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UNITED STATES
DEPARTMENT OF THE INTERIOR
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BUREAU OF MINES
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INFORMATION CIRCULAR

MILLING METHODS AND COSTS AT THE MOUNT ISA MINES, LTD., MOUNT ISA, QUEENSLAND, AUSTRALIA

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BY

J. KRUTTSCHNITT, L. K. JACOBSEN, AND K. B. GROSS

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MILLING METHODS AND COSTS AT THE MOUNT ISA MINES, LTD., MOUNT ISA, QUEENSLAND, AUSTRALIA[1]

By J. Kruttschnitt,[2] L. K. Jacobsen,[3] and K. B. Gross[4]

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[1] The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from Bureau of Mines Information Circular 7073."

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INTRODUCTION

This paper, describing milling practice at Mount Isa Mines, is one of a series covering milling methods and costs that is being published by the Bureau of Mines, United States Department of the Interior.

The Mount Isa concentrator began operations in May 1931 by treating a mixture of oxidized and sulfide ores. Since March 1935 only sulfide ores have been milled; this change involved major alterations to the flow sheet. The maximum capacity on mixed oxide and sulfide ores was 2,650 tons per day; the present capacity is approximately 1,880 tons of sulfide ore per day. The installation of additional crushing, grinding, and flotation equipment when completed early in 1939 will increase the maximum capacity of the concentrator to 2,240 tons of sulfide ore daily.

All tonnages in this paper are expressed in tons of 2,000 pounds, gallon measurements in terms of Imperial gallons of 277.41 cubic inches, and costs in dollars converted at the rate of $4 United States currency to the Australian pound.

ACKNOWLEDGMENT

The authors desire to acknowledge the courtesy extended by the American Cyanamid Co. and Professor Grondijs in permitting the reproduction of the photomicrographs.

SITUATION

The mine, mill, and lead smelter of Mount Isa Mines, Ltd., are situated in northwestern Queensland, 603 miles, by the Government-owned 3-foot, 6-inch-gage railroad, from the port of Townsville, Queensland. The altitude is approximately 1,300 feet above sea level, the climate semiarid, and the average annual rainfall for the district is 16 inches, most of which occurs in the summer.

WATER SUPPLY

Water for milling purposes is obtained from the mine, where an abundant supply is available. This is supplemented when necessary by water from the Rifle Creek reservoir, situated 20 miles from Mount Isa and having a capacity of 1,400,000,000 gallons.

POWER

Electric power, at 3,300 volts, 3-phase, 50 cycles, is supplied from the main power plant, which consists of 3 units. Each unit comprises one Babcock & Wilcox boiler fired with pulverized coal supplying steam at 300 pounds pressure and 750° F. superheat to one 5,000-kilowatt turbo-alternator. Cost of power distributed averaged 0.67 cent per kilowatt-hour for the year ended June 30, 1938. The coal used averaged 14.5 percent ash and 12,600 B.t.u. per pound.

All motors of 75 horsepower and upward are wound for 3,300 volts, the rest being on low-tension 440-volt circuits.

MINING METHODS

Carbonate ores were extracted by open-cut mining and sublevel and flat-back filled stopes, with square-set stopes in localized areas. Sulfide ores are extracted by a system of sublevel stoping with ring drilling and blasting developed to meet the problem presented by the Mount Isa ore bodies to best advantage. These methods are described fully in Information Circular 6978[5/].

[5/] Kruttschnitt, J., and Mann, V. I., Mining Methods and Costs at the Mount Isa Mines, Ltd., Mount Isa, Queensland, Australia: Bureau of Mines Inf. Circ. 6978, 1937, 35 pp.

ORES TREATED

Oxidized Ore

The oxidized ores originally treated were complex in character, the principal lead mineral being cerussite, with anglesite and pyromorphite in small amounts. The zinc content of these ores was low.

It was found that efficient sintering and smelting of the carbonate concentrates depended on the presence of sulfides in the concentrates, particularly pyrite. These sulfides supplied the fuel requirements of the sinter charge and could not be supplanted by fuel such as coke breeze, from extraneous sources. Sulfide ores were milled admixed with the oxidized ores in the ratio of 1 part sulfides to 3 parts oxidized ore to supply the required fuel to produce a satisfactory sinter for blast-furnace operations. This metallurgical necessity had the virtue of absorbing in milling operations ore produced from development in the sulfide zone that otherwise would have been stock-piled.

The mill feed throughout the oxidized ore treatment program, therefore, contained lead as cerussite, galena, and lesser amounts of anglesite and pyromorphite, zinc as smithsonite and sphalerite, iron as pyrite, pyrrhotite, and limonite, a typical assay being as follows:

	Ounces per ton
Ag	5.3
	Percent
Pb	10.5
Zn	4.4
Fe	9.9
SiO_2	52.5
S	4.5

The oxidized ores, when dry, were soft and nonabrasive but tended to pack on the screens and in the Symons crushers when wet, because of the clay content of the altered shale gangue.

The presence of hard chert bands in the carbonate ores made grinding difficult and concentration of oversize chert fragments overloaded the ball mills. The release of these overloads through the trunnion discharge often upset the grinding circuit.

Fairly complete oxidization of the ore deposit extends to a depth of 175 to 200 feet. Immediately beneath the oxidized ore is a zone 50 feet thick, locally termed the "transition zone"; it is characterized by partial oxidation of the sulfides, some secondary enrichment, and a high proportion of water-soluble salts. Ore from this source presents difficulties in concentration because it exerts an adverse effect on the selective lead-zinc separation and metal recoveries. As a result, a

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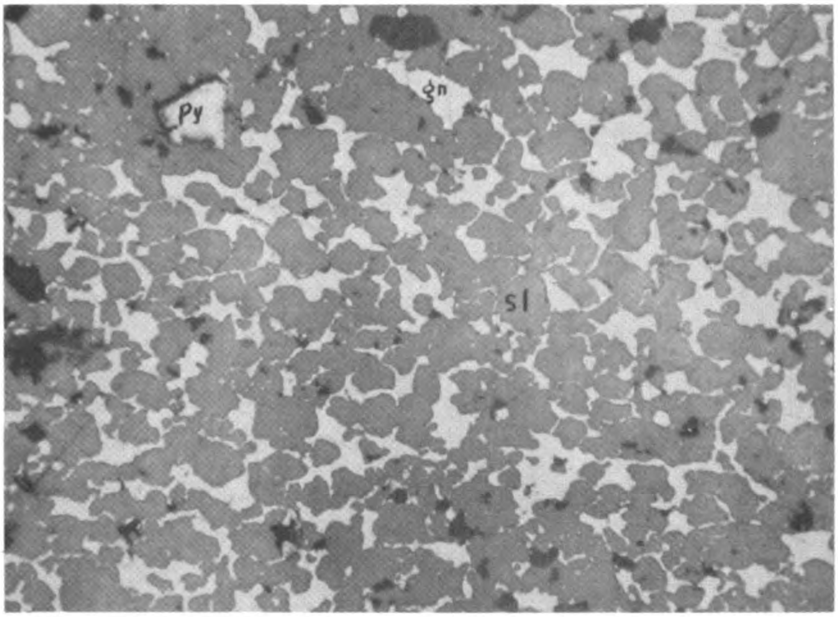

Figure 1.- Intimate mixture of galena and sphalerite with a small amount of pyrite. (Magnification, X55.)

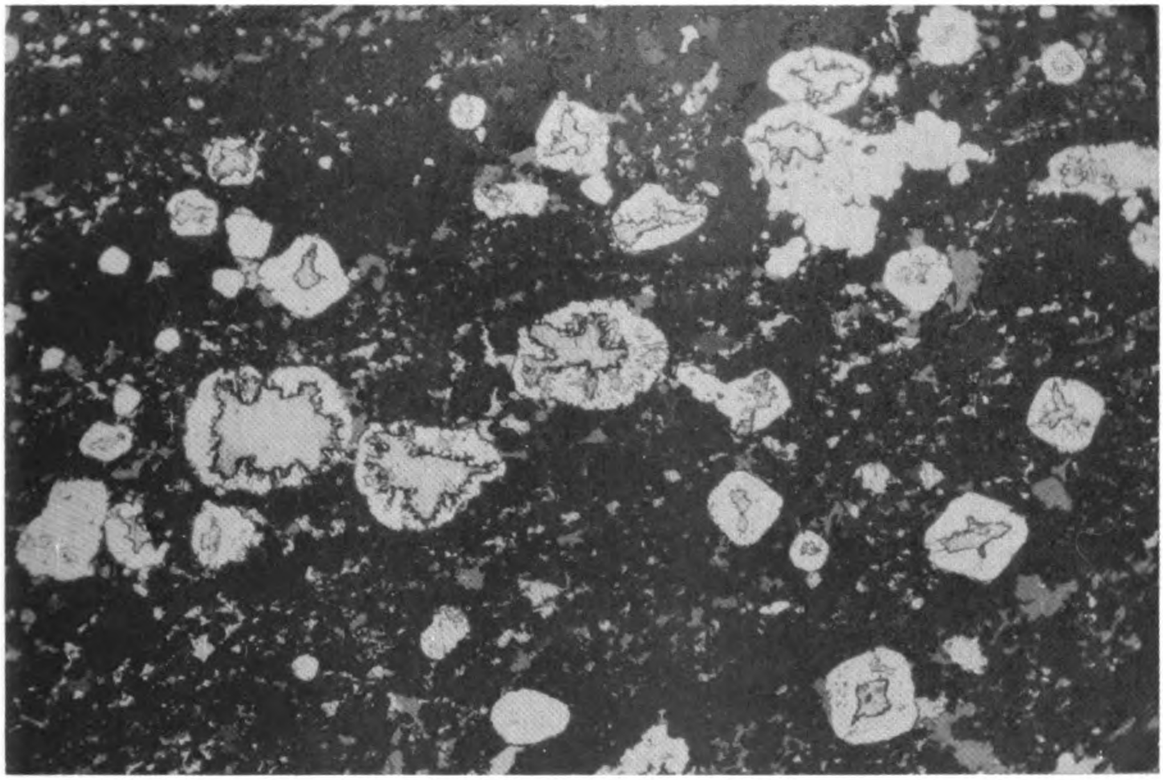

Figure 2.- Pyrite (white) enclosing galena (light gray). The dark areas are gangue. (Magnification, X200.)

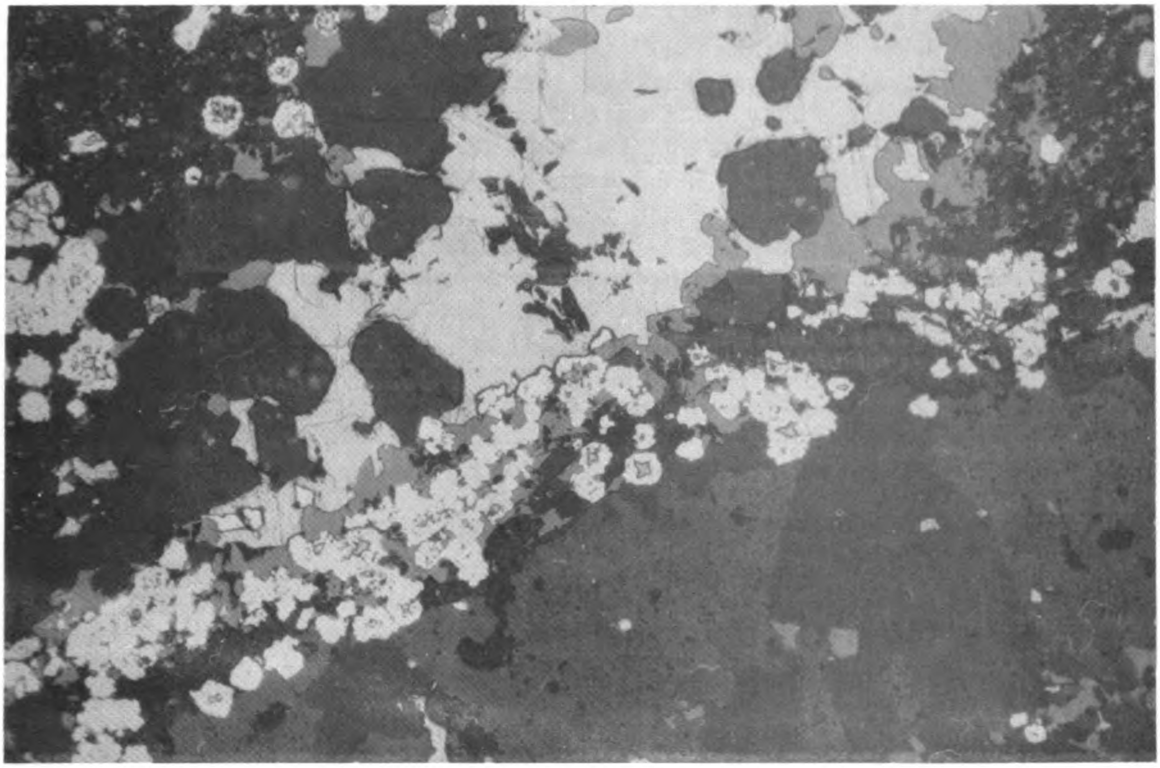

Figure 3.- Pyrite (white) enclosing galena (light gray). The large, dark areas are gangue. Sphalerite (medium gray) borders a large area of galena, with smaller areas of chalcopyrite showing as very slightly darker areas within the galena. (Magnification, X100.)

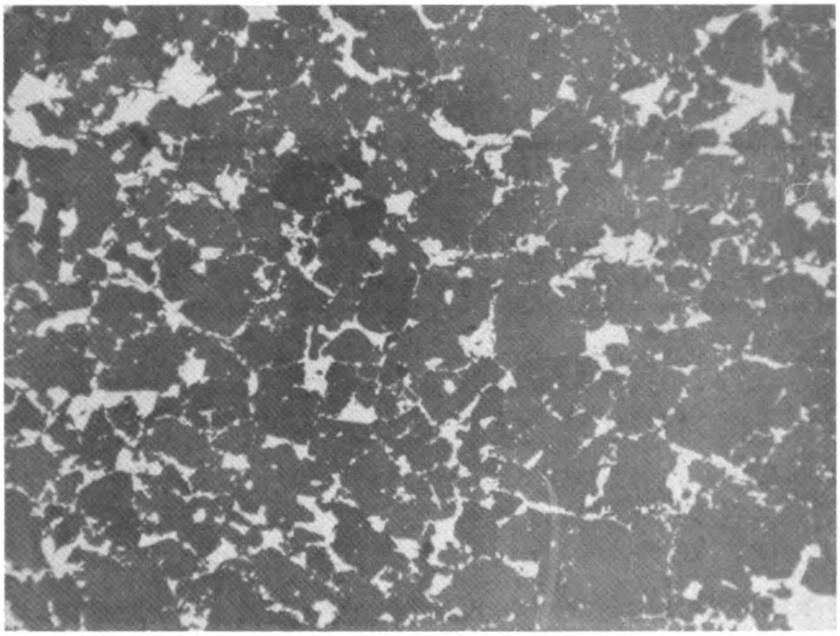

Figure 4.- Galena replacing fine-grained carbonates. Replacement starting at the grain boundaries. (Magnification, X55.)

large percentage of the ore in this zone is being left as a level pillar. Altered ore, however, is found along fissures and cracks, at levels well below the transition zone, but the alteration is not great enough to affect mill metallurgy adversely for long periods.

Sulfide Ore

The sulfide ore now treated consists of a very intimate mixture of galena, sphalerite, pyrite, and pyrrhotite, with smaller amounts of chalcopyrite, marcasite, and complex sulphosalts of silver. The gangue consists chiefly of dolomitized, stratified shale with some silica and carboniferous material, principally graphite.

Concentration is made difficult by the fineness of the mineralization, and by the intimate degree of association of the sulfide minerals and of the sulfide and gangue minerals. Three typical occurrences are worthy of special mention and are shown in the illustrations:

1. Bands up to several inches in thickness, consisting wholly of an intimate mixture of galena and sphalerite with small amounts of pyrrhotite and pyrite and very little gangue. This structure is prominent in ore from the deeper levels of the Black Star ore body and is illustrated in figure 1.

2. Pyrite, partly replaced by galena to give in many instances a curious atoll-like structure in cross section (see figs. 2 and 3). This structure is prominently represented in ore from the Black Star ore body.

3. Gangue minerals, partly replaced by galena, giving an extremely fine association (fig. 4). This structure is prominent in ore from the Rio Grande ore body.

Ores from the Black Star lode are characterized by their lower grade, extremely fine texture, and the presence of much pyrite in otherwise barren bands of shale. This ore body represents 97 percent of the developed reserves.

Ores from the Black Rock and Rio Grande ore bodies are of higher grade, and much of the galena occurs in a relatively coarse state. Ores containing less than 10 percent lead are classed as low-grade and 10 percent or more lead as high-grade.

The sulfide ore is only moderately abrasive, but its resistance to crushing and grinding is high. It breaks preferentially along bedding planes, and consequently a difficult crushing and screening problem is reflected in the design of the crushing plant. The proportion of primary fines is low - generally less than 10 percent through a 1/4 inch

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aperture - and the ore is seldom wet enough to give any screening troubles, the average moisture in the mill heads being only 1.2 percent.

Largely due to the high proportion of finely divided pyrite, pyrrhotite, and minor quantities of marcasite the ore rapidly oxidizes if allowed to remain broken in the stopes. This factor was considered in selecting a method of mining that would result in a very short time lag between ore breaking and milling.

For a most informative description of the mineralization and structure of the Mount Isa ore bodies, the reader is referred to articles by H. F. Grondjs and C. Schouten in Economic Geology, June-July 1937,[6] and R. Blanchard and G. Hall in Economic Geology, December 1937.[7]

BRIEF HISTORY OF CONCENTRATOR OPERATIONS

The concentrator and crushing plant are situated on a hillside adjacent to the main ore-hoisting shaft of the mine, allowing, as far as possible, for gravity flow of ore products through the mill.

The buildings are of steel construction provided with crane runways over the crushers, ball mills, classifiers, and flotation sections. Floors are concrete and wood.

As originally designed, the mill was arranged for the separate treatment of oxidized and sulfide ores in two oxide units of 830 tons capacity each and one sulfide unit of 560 tons capacity[8] but when production began in May 1931 the two oxide units, for reasons already explained, began treating a mixture of oxide and sulfide ores in the proportion of 3 to 1, respectively. The sulfide unit was converted to treat mixed ores.

A detailed description of these operations has been published.[9] The flow sheet shown in figure 5 indicates the general scheme of treatment, the daily capacity being 1,850 tons. In 1933 the capacity of the mill was raised to 2,650 tons by the addition of three duplex five-compartment May jigs and six additional Deister Plat-O tables. Flotation capacity also was increased by the addition of two 45-foot Forrester cells. This flow sheet is shown in figure 6.

By March 1935 reserves of carbonate ore were nearly exhausted and a change to treat straight sulfide ore became necessary. The harder, compact, intimately associated sulfide ore necessitated many changes in the flow sheet, the principal ones being:

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[6] A study of the Mount Isa Ores by H. F. Grondijs and C. Schouten: Econ. Geol., vol. 32, No. 4, June-July 1937.

[7] Mount Isa Ore Deposition, by R. Blanchard and G. Hall: Econ. Geol., vol. 32, No. 8, December 1937.

[8] See Mining Magazine, vol. 41, No. 6, December 1929.

[9] L. K. Jacobsen, Milling Methods and Costs at the Concentrator of Mount Isa Mines, Ltd.: Australasian Inst. Min. and Met., New Ser. No. 90, 1933.

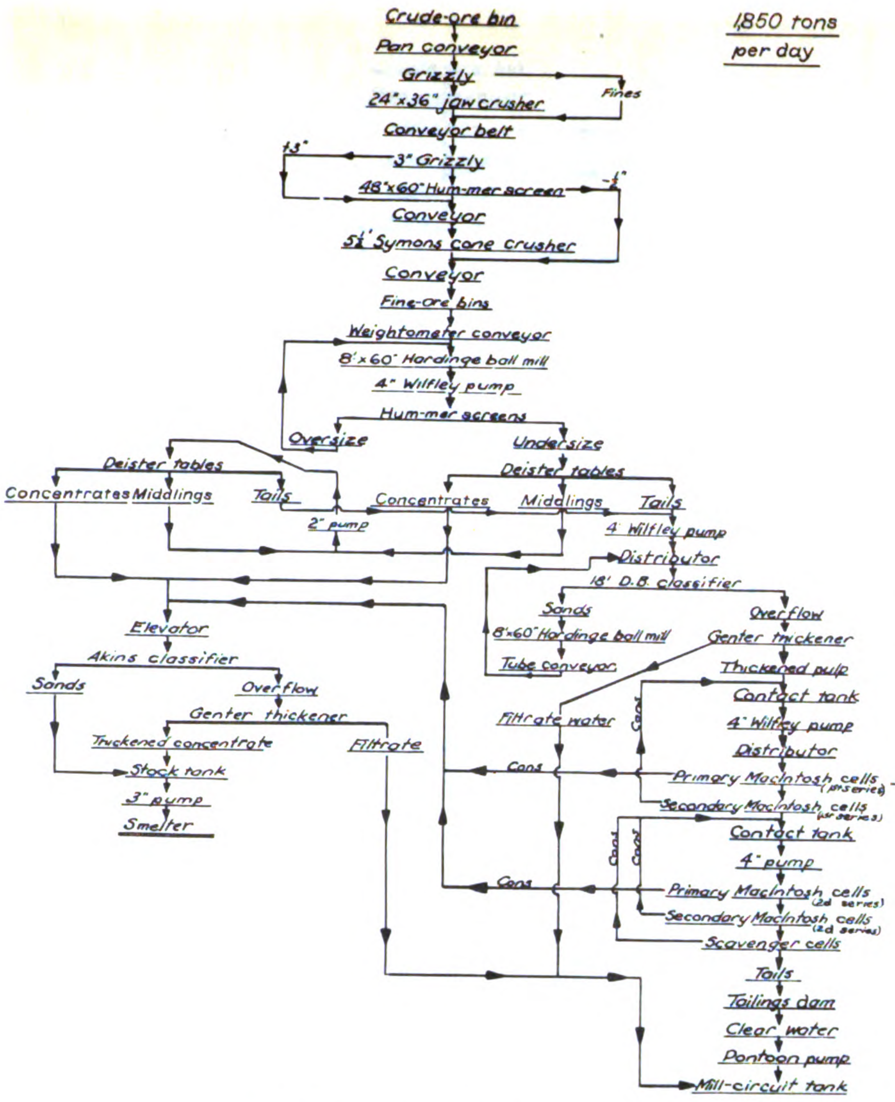

Figure 5.- Carbonate-sulphide flow sheet.

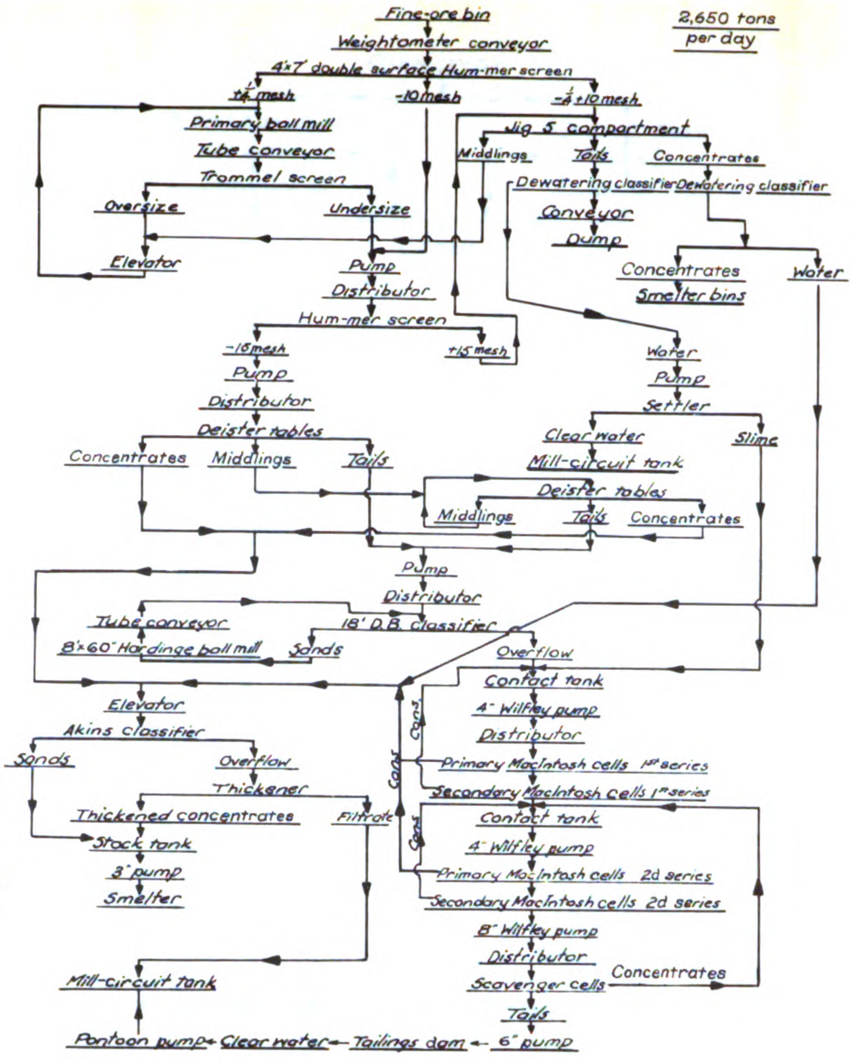

Figure 6.- Carbonate-sulphide flow sheet.

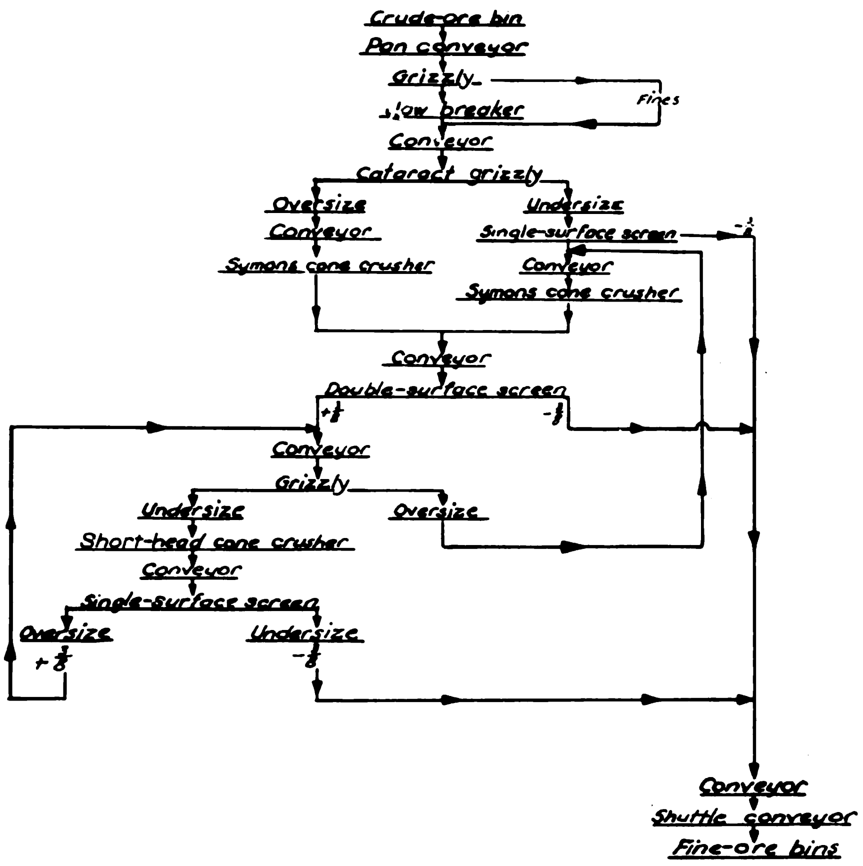

Figure 7.- Crushing-plant flow sheet.

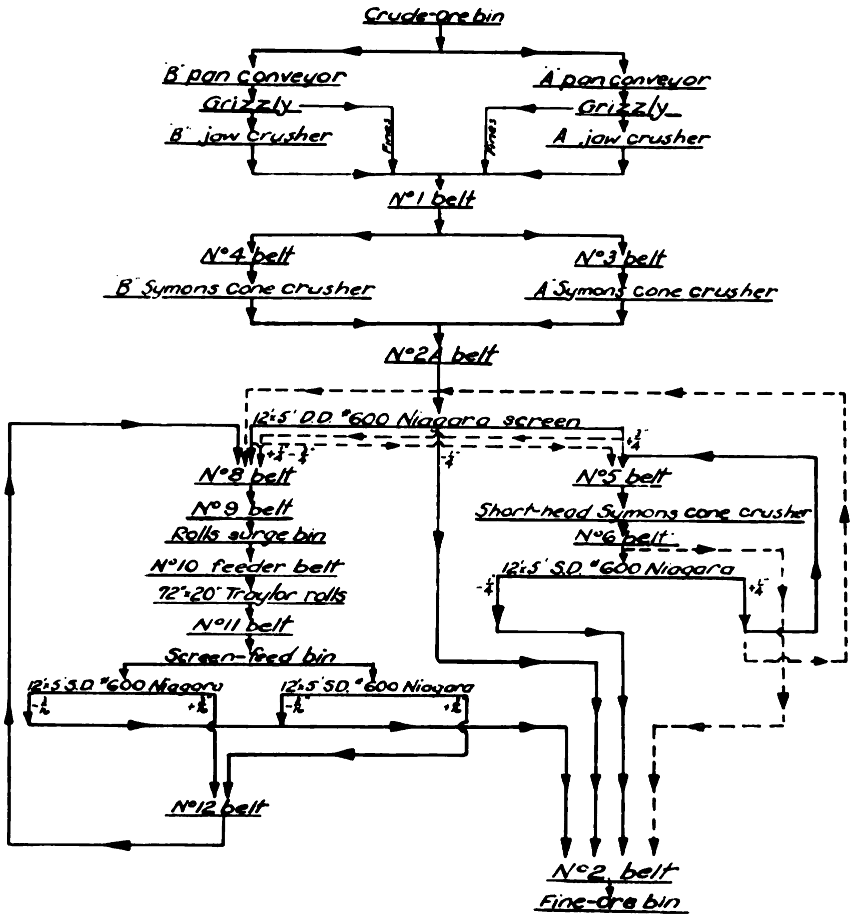

Figure 8.- Crushing-plant flow sheet, including Traylor rolls.

1. Elimination of gravity concentration (jigs and tables).
2. Provision of an additional stage of crushing.
3. Provision of classifiers in the primary grinding circuit.
4. Installation of equipment for the recovery of zinc concentrates.

These changes were made without seriously interfering with production and were completed by October 1936.

As mining has progressed to deeper levels, a decided tendency to closer association of the sulfide minerals has been observed together with a definite increase in resistance to crushing and grinding. To maintain output and improve metal recoveries increased comminution became necessary; the three following major changes in flow sheet were made:

1. Existing bowl classifiers in the secondary circuit (Dorr Model D 8 feet wide by 18-foot bowls) were replaced with classifiers of modern design and larger capacity (Dorr model QSFXB 16 feet wide by 20-foot bowls).

2. A tertiary stage of grinding was added, comprising an additional ball mill and the three bowl classifiers formerly in the secondary circuit.

3. The crushing plant was equipped with screens of larger capacity and the flow sheet was revised.

These changes were completed in 1937. More recently (July 1938) all of the existing lead-flotation equipment was replaced by Fagergren machines.

Additional crushing-plant capacity and an improvement in the fineness of product will ensue when heavy-duty rolls (72- by 20-inch) at present under construction are completed.

The present concentrator capacity of 1,880 tons is being increased to a maximum of 2,240 tons daily by the addition of a 10-foot by 72-inch Hardinge ball mill and additional flotation equipment.

Table 1 shows the improvement in lead metallurgy that has resulted from the introduction of the improvements listed above and also from the decline in the proportion of altered ore from the vicinity of the "transition zone."

PRESENT CONCENTRATOR METHODS

Crushing

The present flow sheet of the crushing plant is shown in figure 7.

All ore delivered to the coarse ore bin has previously passed through grizzlies with 14- by 20-inch openings located at various points underground.

The coarse-ore bin has a capacity of 1,600 tons, of which approximately 1,100 tons is "live" storage.

From the coarse-ore bin the ore is fed to two 36- by 24-inch Blake-type Fraser and Chalmers jaw breakers by two 36-inch apron feeders. The jaw breakers are set to 4 inches and are preceded by fixed bar grizzlies set at 2-1/4 inches. This close grizzly setting is necessitated by the pronounced tendency of the ore to break in flat slabs along bedding planes. Even at these settings enough flat slabs pass the jaw breakers and grizzlies to justify the employment of a man who spalls any rocks large enough to cause hanging up in the chutes. The apron feeders are driven by 10-horsepower motors through variable-speed drives, providing regulation in feed rate. The jaw breakers are belt driven at 224 r.p.m. by 75-horsepower motors.

The jaws of the breakers are lined with manganese steel, vertically corrugated, the life of the various sections being as follows:

	Months
Fixed jaw, upper section.....	20
Fixed jaw, lower section.....	12
Movable jaw, upper section...	20
Movable jaw, lower section...	12

Toggles last approximately 16 weeks, and toggle seats 16 weeks, but recent improvements in design and lubrication are expected to extend the life of these parts.

The combined jaw-breaker product and grizzly undersize discharge onto a 30-inch conveyor belt inclined at 19-1/2°, it feeds to a fine-roll cataract grizzly set at 2 inches. The oversize product from the cataract grizzly is fed by a 36-inch conveyor to one 5-1/2-foot Symons crusher set at 3/8 inch. The primary undersize through the grizzly consisting of less than 10 percent of the product is screened by a 12- by 5-foot Tyler Niagara single-surface screen having 3/8- by 3/8-inch apertures. A 36-inch belt conveyor delivers the screen oversize to a second 5-1/2-foot Symons crusher set at 3/8 inch.

Both Symons crushers are driven by V-ropes at 485 r.p.m. by 150-horsepower motors and are protected against tramp iron by two 2-pole Dings high-intensity magnets. The cataract grizzly is driven through a worm speed-reducer and chain drive by a 15-horsepower motor; the Niagara screen is V-rope driven at 960 r.p.m. by a 7-1/2-horsepower motor.

The combined product of both 5-1/2-foot Symons crushers is delivered by a 30-inch belt conveyor, inclined at 23°, to one double-surface Tyler Niagara screen. This screen is 12 by 5 feet and is V-rope driven at 960 r.p.m. by a 7-1/2-horsepower motor. The top surface apertures are 3/4 by 3/4-inch; the lower ones are 3/4-by 1/4-inch.

Although the Symons crushers are set at 3/8 inch, the product contains occasional flat fragments up to 5 inches long, 4 inches wide, and about 1/2 inch thick. For this reason the oversize from both screen surfaces of the Niagara screen is delivered to a fixed bar grizzly set at 1 inch and superimposed above the 4-foot short-head Symons crusher. The oversize from this grizzly is returned to one of the 5-1/2-foot Symons. The undersize comprises the feed to the short-head Symons; it is set at 1/8 inch and operates in closed circuit with an 8- by 3-foot Niagara single-surface screen with 3/8- by 3/8-inch apertures. This screen is V-rope driven at 960 r.p.m. by a 5-horsepower motor. The short-head Symons is tex-rope driven at 485 r.p.m. by a 100-horsepower motor. Magnetic protection also is provided for this crusher.

At present (December 1938) the capacity of the short-head Symons limits the throughput to 140 tons per hour, at which rate only one jaw breaker is run.

Undersize from all three Niagara screens discharges onto a 30-inch conveyor and is delivered at the top of the fine-ore bins onto a shuttle conveyor under which there are six cylindrical steel bins 22 feet in diameter by 45 feet high. Even distribution of the ore is achieved by means of a traversing gear that moves the shuttle to each bin in turn. The total live storage is approximately 5,500 tons.

Bowl and mantle liners of the three Symons crushers are manganese steel. Present practice is to grind worn faces to correct profile during the life of the liners to insure more efficient performances. Liner lives are set out below:

		Days
5-1/2-foot Symons	- Mantle liner.....	350
	Bowl liner.......	350
4-foot short-head Symons	- Mantle liner.....	100-114
	Bowl liner.......	100-114

The Niagara screen coverings are woven from manganese-steel wire, the respective lives being as follows:

	Days
12- by 5-foot single-deck Niagara.....	18
12- by 5-foot double-deck Niagara:	
Upper screen.......	42
Lower screen.......	60

Power consumed by the jaw breakers and large Symons crushers fluctuates widely but averages as follows:

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	Horsepower
Jaw breakers........................	50-60
5-1/2 foot Symons..................	100
4-foot short-head Symons (steady)...	100

The characteristic breaking of the ore along bedding planes persists throughout the range of crushing. The ore also has an extremely high crushing resistance and tends to pack under impact, even when dry, so that high circulating loads are built up in closed-circuit crushing units, and packing produces severe pounding stresses in the Symons crushers when set to deliver a fine product (less than 1/4 inch). With the twofold object of increasing capacity and producing a finer crushed product than is at present possible, heavy-duty, high-speed rolls are now being installed.

The flow sheet (fig. 8) shows how it is proposed to introduce the new equipment into the existing plant. The rolls are Traylor heavy-duty, 4-tension, rod-type, 72 inches in diameter by 20-inch face and will be driven through V ropes by two 125-horsepower motors at 115 r.p.m. The alternative routes to provide flexibility are shown by broken lines in figure 8.

In the revised flow sheet, the Symons crushers will be operating over a range of reduction where they have proved efficient on this ore, and packing and pounding should be eliminated.

Dust-handling system

Incorporated in the new crushing-plant flow sheet is a dust-recovery system composed of two 17,000-cubic foot fans and two Western Precipitation Co. Multiclone dust collectors type 9VII, size 50.

Grinding

Until March 1937, each of the three grinding units comprised two 8-foot by 60-inch Hardinge ball mills grinding in two stages; the primary mill was in closed circuit with an 8-foot model F Dorr classifier, whereas the secondary mill was in closed circuit with a model D, 8-foot wide by 18-foot bowl Dorr classifier.

During March and April 1937 the three model D classifiers serving the secondary ball mills were replaced with model QSFXB Dorr classifiers 16 feet wide by 20-foot bowl in order to cope with the large circulating loads required to improve stage grinding.

The three model D classifiers were relocated in a new tertiary regrind circuit coupled with a new 8-foot by 60-inch Hardinge ball mill. This addition was completed in October 1937.

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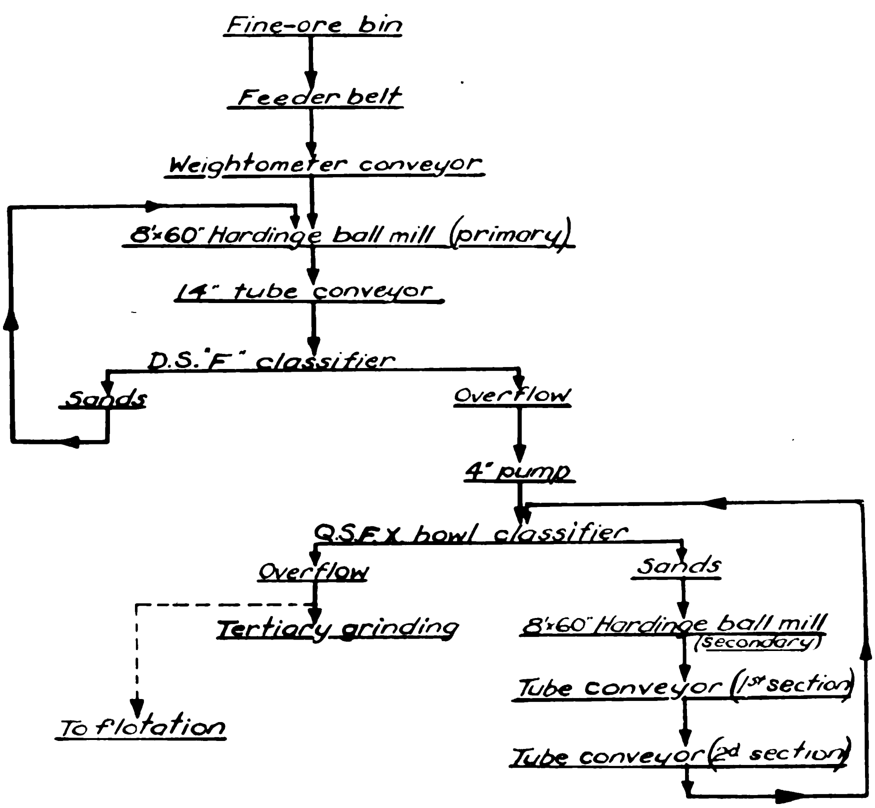

Figure 9.- Grinding unit, two-stage flow sheet.

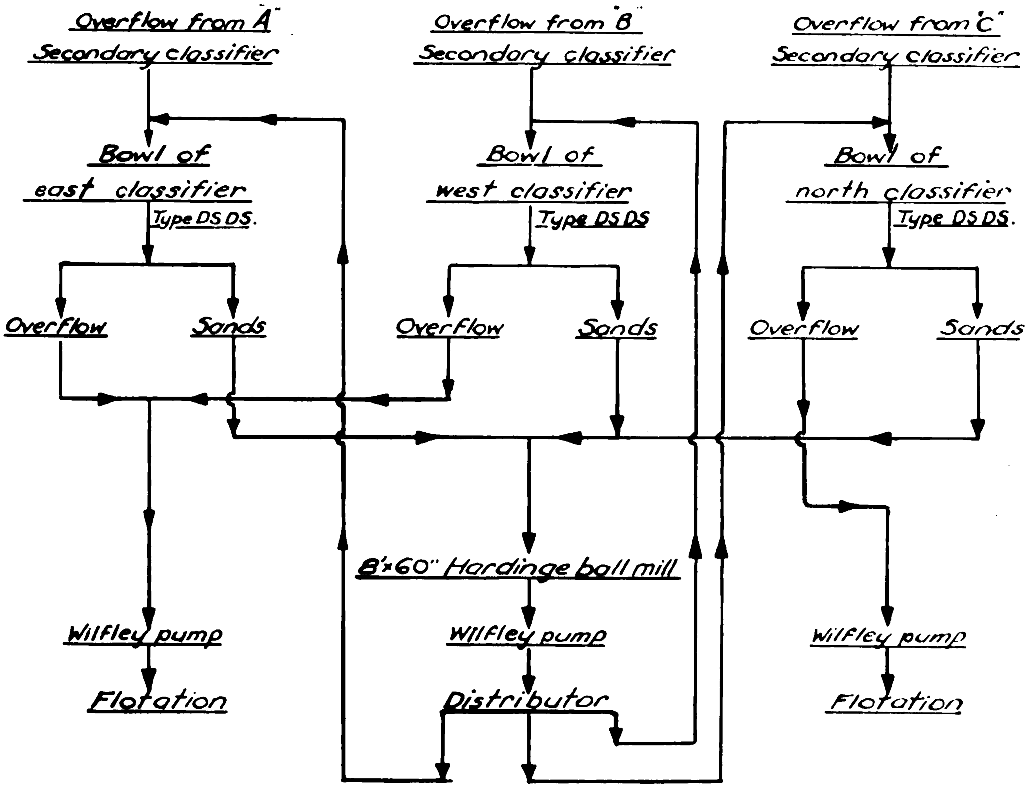

Figure 10.- Regrind or tertiary grinding flow sheet.

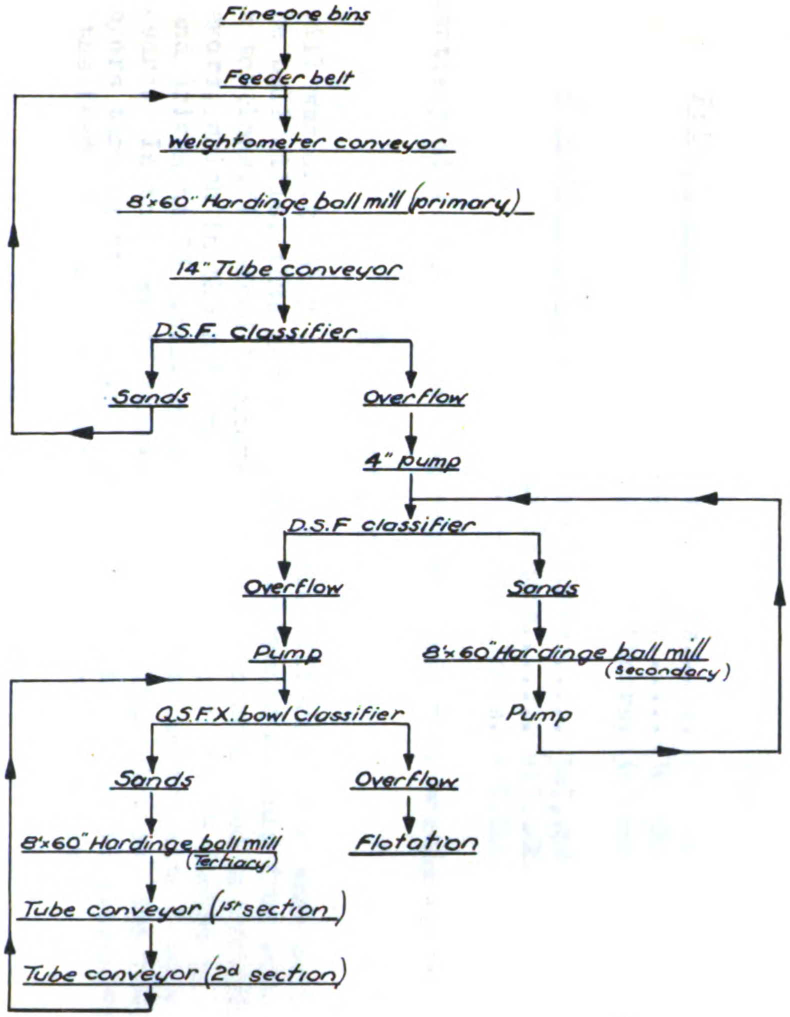

Figure 11.- Grinding unit, three-stage flow sheet.

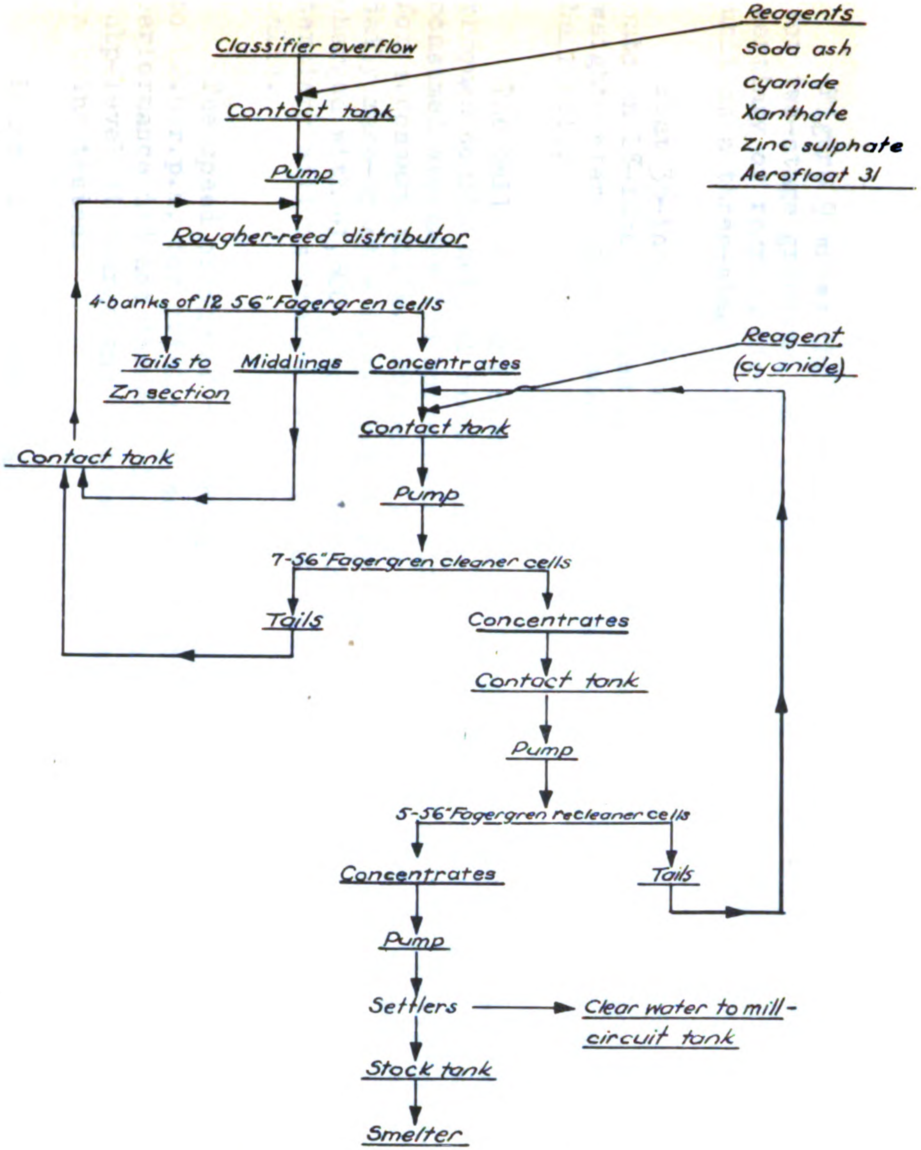

Figure 12.- Lead flotation flow sheet.

Figure 9 shows the flow sheet of one unit as at present operated for two-stage grinding, and figure 10 shows the flow sheet of the tertiary or regrind circuit. Figure 11 shows the flow sheet of one unit as a three-stage grinding circuit.

Flat 36-inch belt conveyors under each fine-ore bin feed in pairs onto an 18-inch inclined conveyor that passes over a Blake-Denison weightometer before discharging into the feed scoop box of a primary ball mill.

The ball mills are operated at 20.8 r.p.m. by 220-horsepower motors through double helical reduction (4 : 1 ratio) and spur gears. Power consumed averages 195 horsepower for primary mills and 205 horsepower for secondary mills. Primary mills carry a load of 44,000 pounds with daily make-up of 4 inch forged chrome-steel balls. Secondary mills are charged with 41,500 pounds of 2 inch alloy cast-iron balls, whereas the tertiary mill is charged with 38,000 pounds of 1-1/2 inch alloy cast-iron balls.

The speed of one of the primary mills was increased from 20.8 r.p.m. to 22.5 r.p.m. for trial purposes, but with the lifter bar liner mill performance did not increase in proportion. The installation of a high pulp-level discharge and reverse cone grate on one of the primary mills is being tested.

Liner life is recorded as follows:

		Tons
Primary mills	- Feed-cone liners......	287,000
	Cylinder liners.......	307,000
	Discharge-cone liners.	367,000
Secondary mills	- Feed-cone liners......	367,000
	Cylinder liners.......	417,000
	Discharge-cone liners.	470,000

No tertiary mill liners have been replaced after 12 months operation.

Pipe Conveyors

The mill was originally designed with classifiers on the same center line as the ball mills, thus necessitating conveyance of pulp from mill discharges to classifier-feed launders. This is accomplished by revolving pipe conveyors, which in the primary grinding circuit are 14 inches in diameter and driven at 38 r.p.m. by 7-1/2-horsepower motors with V ropes and spur gears. In the case of the secondary grinding circuit, two 14-inch pipe conveyors turning at 31 r.p.m. operate in series to avoid excessive sizes for the feed scoops. Feed-scoop diameters are as follows:

	Radius, inches
Primary ball mills.............	42
Primary pipe conveyor..........	29
Secondary ball mill............	54
Secondary pipe conveyor........	36

All scoops are fitted with renewable manganese steel shoes.

Classification

Primary Classifiers

The primary classifiers are model F Dorr units 8 feet wide, sloping 2-1/2 inches per foot and driven at 28 strokes per minute by 15-horsepower motors. Primary ball-mill discharge entering these classifiers has an average density of 83 percent solids with the overflow normally at 70-percent solids.

For Mount Isa sulfide ore crushed to 90 percent minus 1/4-inch, the maximum circulating load of the primary grinding circuit established by tests over an extended period is 70 tons total feed per hour, of which 27 tons represents new feed. When this maximum is exceeded overloads build up, and conditions in the circuit become unbalanced.

Secondary Classifiers

The overflow of the primary classifiers, with an average density of 70 percent solids, is pumped by a 4-inch Wilfley pump to the secondary Dorr QSFX bowl classifiers, the tanks of which are 16 feet wide by 36 feet long and slope at 2 inches per foot. The four reciprocating rakes are driven at 17 strokes per minute by a 25-horsepower motor, but actual power required under existing conditions is 18 horsepower. Rakes in the 20-foot bowls operate at 3.3 r.p.m. The overflow density ranges from 38 to 48 percent solids, depending on the tonnage and type of ore, but averages 42 percent solids.

High circulating loads are customary in the secondary grinding circuits, reaching, on the average, 200 to 220 tons per hour of total feed to each mill.

Tertiary Classifiers

The overflows from the three secondary bowl classifiers are distributed to three model D classifiers of the tertiary circuit. These model D units, previously employed in the secondary grinding circuit, have 8-foot-wide raking compartments and 18-foot bowls. They operate at 16 strokes per minute, and the bowl rakes operate at 2 r.p.m.

1. 10-foot by 72-inch ball mill.
2. Dorr multizone classifier.
3. 450-horsepower induction motor.
4. Weightmeter, conveyor D.
5. Weightmeter, conveyor A.
6. Weightmeter, conveyor B.
7. Weightmeter, conveyor C.
8. 30-inch shuttle conveyor.
9. 30-inch feeder conveyor to conveyor D.
10. 30-inch feeder conveyors to conveyors A and B.
11. 30-inch feeder conveyor to conveyor C.
12. Model F Dorr duplex classifier.
13. QSFXB Dorr duplex classifier.
14. 4-inch Wilfley pumps.
15. 3-inch Wilfley pump to smelter.
16. Lead stock tank under floor.
17. Settlers - lead concentrate thickening.
18. Three-way distributors.
19. Surge tank.
20. Soda-ash tank.
21. Lift.
22. Tertiary grinding section, 3 model D Dorr classifier, 1 DSF Dorr classifier, 1 8-foot by 60-inch Hardinge ball mill (shown in dotted lines).
23. Lead contact tanks.
24. Two-way distributor.
25. Four-way distributor.
26. Five Fagergren recleaner cells, 56- by 14-inch.
27. Seven Fagergren cleaner cells, 56-inch square type.
28. Twelve Fagergren rougher cells, 56-inch square type.
29. Six Fagergren roughers, 56-inch round type.
30. Five Fagergren roughers, 56-inch square type.
31. 14-foot-diameter stock tank.
32. 3-inch Wilfley pump to zinc filter tank.
33. 30- by 10-foot Dorr thickener.
34. Three 24-inch Fahrenwald recleaner cells.
35. Four 24-inch Fahrenwald recleaner cells.
36. Five 24-inch Fahrenwald recleaner cells.
37. Distributor.
38. Zinc contact tanks.
39. 24-inch Fahrenwald roughers.
40. Surge tank.

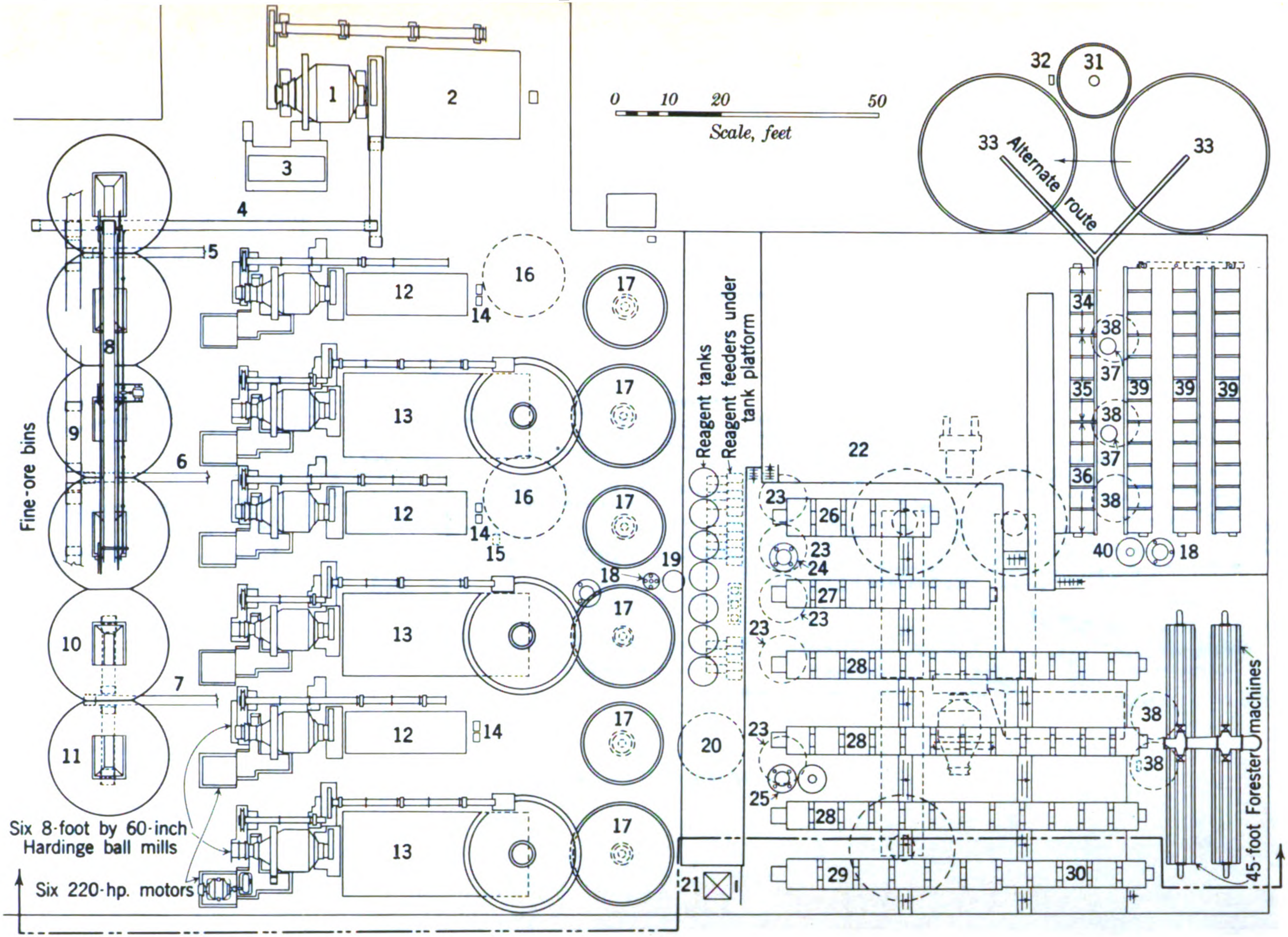

Figure 13.—Plan and general arrangement, Mt. Isa concentrator.

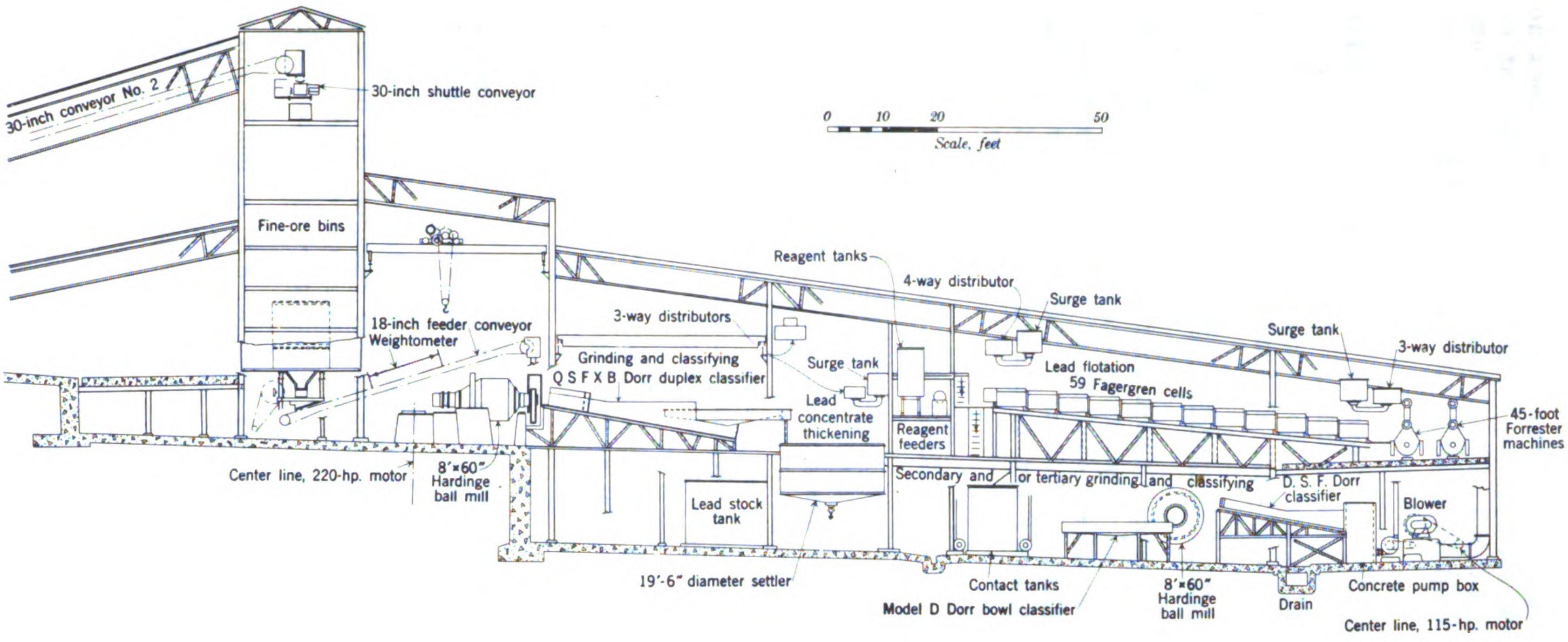

Figure 14.—Section *A–A*, Mt. Isa concentrator.

Flotation feed comprises the overflow of the three tertiary classifiers and goes to the cells at 33 to 38 percent solids. Sands from these classifiers are reground in an 8-foot by 60-inch Hardinge ball mill. The mill discharge, at 77 percent solids, is distributed equally to the three classifiers.

By operating these classifiers to overflow a high-density pulp, differential grinding of the heavy sulfide middlings is attained, so that, although 76 percent of the flotation feed is minus-325 mesh (Tyler), over 90 percent of the galena is contained in this fraction.

Two-Stage versus Three-Stage Grinding

Results to date indicate that on Mount Isa sulfide ores better metallurgy results from three-stage grinding. It is proposed, however, to include a new primary 10-foot by 72-inch Hardinge ball mill and a 16-foot-wide QFX multizone classifier in the grinding circuit to facilitate further testing of the relative merits of two-stage and three-stage grinding of these ores.

Lead Flotation

Present lead-flotation equipment consists of 60 Fagergren cells, 6 of which are 56-inch round cells, and the rest are 56-inch square cells arranged in parallel rows; the installation was completed in July 1938. The rougher section consists of four rows of 12 cells each. The cleaner and recleaner rows contain 7 and 5 square cells, respectively. The flow sheet is shown in figure 12. The installation presents an accessible and compact unit. Figures 13 and 14 show the plan and side elevation.

The rougher cells are driven at 580 r.p.m. by 10-horsepower motors. Cleaner and recleaner cells are operated at 400 r.p.m., tex-rope driven by 7-1/2-horsepower motors. Rotor and stator wearing parts are rubber covered.

The Fagergren cell was adopted for the new installation after extensive trials with a unit consisting of six 56-inch round cells.

The tertiary bowl classifier overflows, which comprise the flotation feed, are pumped to two contact tanks, where they are mixed with the rougher middlings and the cleaner tailings; this product is then delivered to the feed distributor by 4-inch Wilfley pumps.

Reagents are added to the contact tanks; attempts to obtain better contact by adding reagents to the ball mills gave poor results. The following reagents were added:

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	Pounds per ton
Soda ash........................	1.25
Aero cyanide (50 percent NaCN)....	.60
Potassium ethyl xanthate..........	.30
Zinc sulfate......................	.60
Aerofloat 31......................	.22

The pH of the flotation feed ranges from 8.0 to 8.6, averaging 8.3

Soluble reagents are fed at 10-percent strength by disk and cup feeders. The Aerofloat is stage-fed to the new feed and the sixth and ninth cells of each row of roughers by multicompartment Clarkson feeders; any individual cell, however, can be stage-oiled if required.

When mine water is added directly to the circuit, it is immediately necessary to increase the reagents, particularly Aerofloat, to hold metallurgical results constant. As a general practice, the make-up mine water is added to the mill tailings and a conditioning action takes place, due to aeration and contact, that renders the return circuit suitable for milling.

The number of cells making middling may be changed readily. At present, five cells in each row make a rougher concentrate, the froth from the remaining 7 cells being returned as a middling to the head of the circuit. It has been found impossible to reduce this heavy circulation of middling without severe loss of lead and silver in the tailings.

Screen analyses and assays of the various products are given in tables 5 and 6 (pp.25 and 26).

Zinc Flotation

The tailing from the lead-flotation plant (zinc-flotation feed) is conditioned in two 9-foot-diameter by 12 foot tanks in series. Copper sulfate and Aerofloat are added to the contact tanks, xanthate and sodium Aerofloat to the feed-pump sump. Pulp is then delivered to the feed distributor of the zinc-flotation plant by an 8-inch Wilfley pump.

The zinc-flotation equipment comprises forty-eight 24-inch Fahrenwald cells arranged in 4 rows of 12 cells. The impellers are driven in pairs by V-rope drives at 238 r.p.m., one 15-horsepower motor to each pair. Scrapers for froth removal are driven at 17 r.p.m. by one 1-horsepower motor to each row of 12 cells.

A rougher concentrate is produced from the first 8 cells in each of three rows; a middling from the last 4 cells of each row is returned to the head of the circuit. The cleaner bank of 12 cells is arranged to provide triple cleaning, there being 5, 4, and 3 cells for the respective operations. A 9-foot-diameter by 12 foot surge tank feeds each stage of cleaning. The following reagents are used:

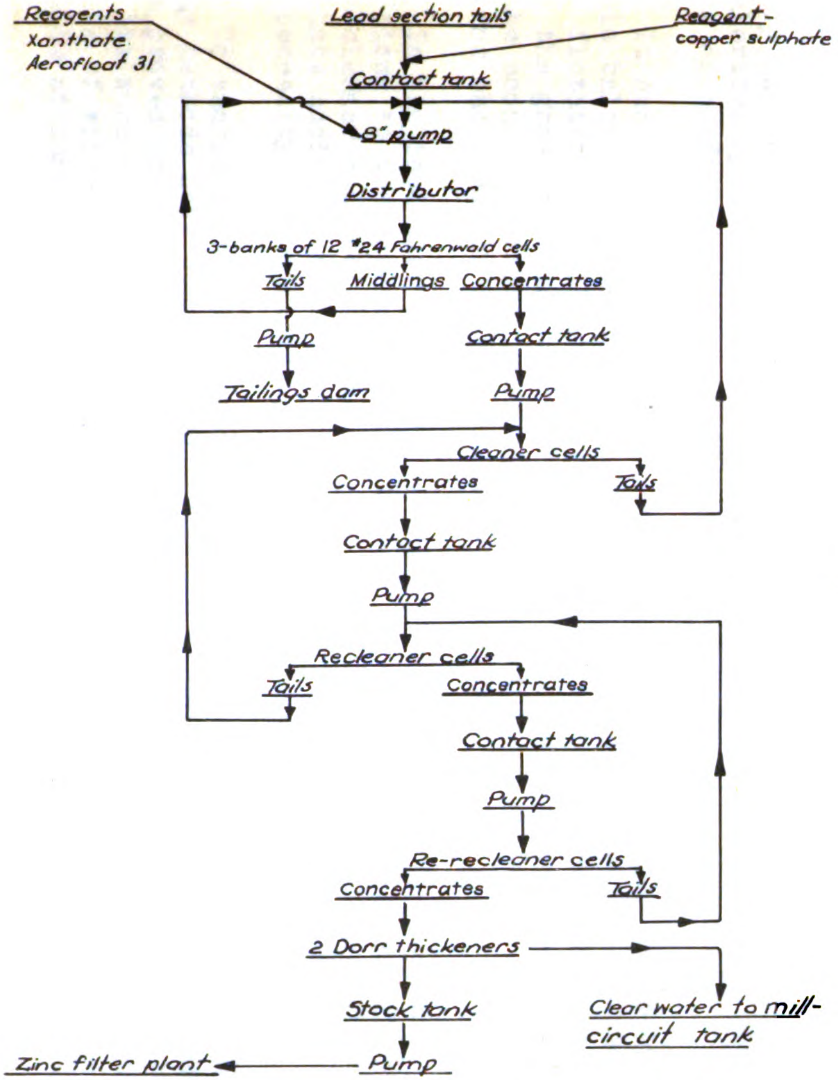

Figure 15.- Zinc-flotation flow sheet.

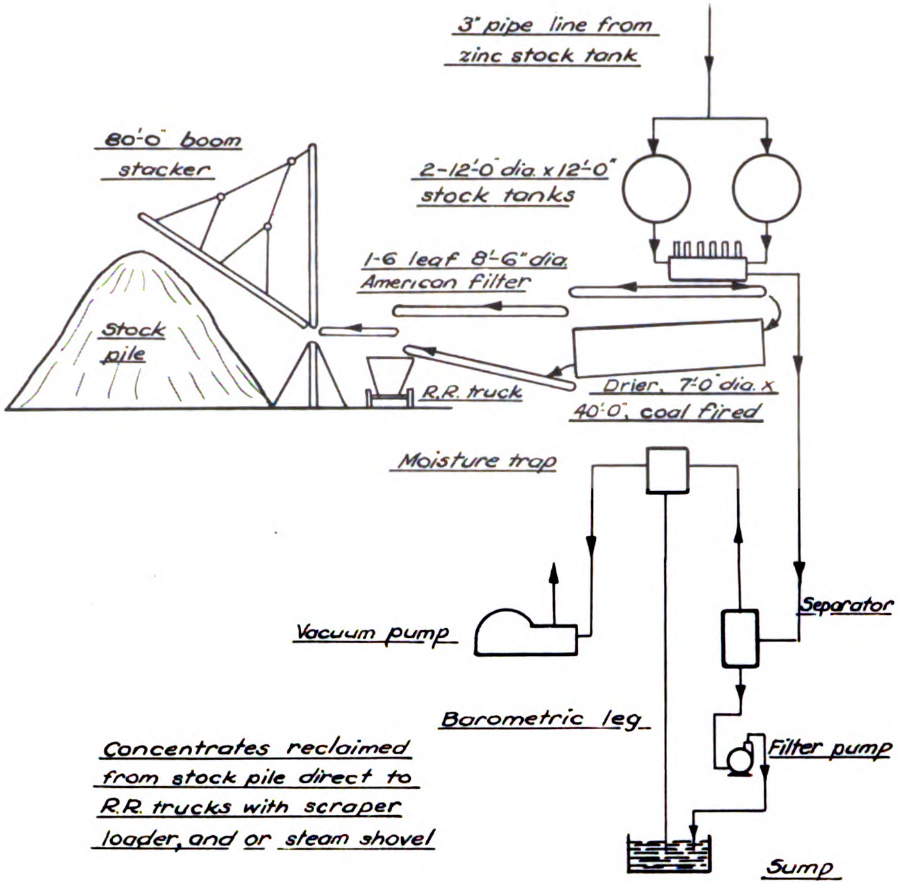

Figure 16.- Zinc filter plant.

	Pounds per ton
Copper sulfate..............	1.045
Potassium ethyl xanthate.....	.076
Sodium Aerofloat............	.083
Aerofloat 31................	.033

Copper sulfate is fed by means of a saturator to which water is delivered by a disk and cup feeder. The strength of the solution is determined at regular intervals by means of the cyanide assay method; the rate of adding water is adjusted by reference to convenient tables.

Xanthate and sodium Aerofloat are fed at 10-percent strength by disk and cup feeder, the Aerofloat 13 being controlled by a Clarkson feeder.

The Fahrenwald cells are now fitted with rubber-covered impellers and hood plates. Hard cast-iron hood plates were replaced in 6 months, cell pans after 12 months, and the baffle castings after 18 months operation. The impellers are giving longer service with rubber-covered hood plates and are estimated to give up to 3 years of life. Flow sheet of the zinc section is shown in figure 15.

Control of Flotation Operations

With the extremely fine and intimate association of lead, zinc, and iron sulfides, conventional methods for gaging tailings losses, such as hand panning or pilot tables, are of doubtful value. The lead lost is mainly in the form of middling grains, common associations being galena-sphalerite, pyrite grains enclosing a core of galena, and various combinations of sulfides and gangue minerals.

It has been found that for a given fineness of grinding and under stable balanced operating conditions the tailing assay in both lead and zinc flotation is indicated correctly by the grade of concentrate produced. Once the grade of concentrate desired has been established by a study of the economic factors involved, the simplest means of control is to aim for the grade of concentrate thus determined.

In the lead circuit, grade is controlled by varying the frother additions to the rougher circuit as well as the rate of froth overflow from cleaner and recleaner cells. In the zinc circuit, however, the xanthate additions are varied, the frother additions being determined by the necessity for maintaining a suitable type of froth.

To enable flotation operators to control concentrate grade, rapid assay methods have been devised for both lead and zinc concentrates and have proved accurate enough for control purposes. The lead assay depends for its success on the fact that the iron is present chiefly as pyrite, little of which is decomposed by the hydrochloric acid used, while the absence of manganese in the zinc concentrate enables the direct method to be employed successfully. Determinations by operators may be completed in 20 to 30 minutes, which includes time employed in drying the sample.

Dewatering and Handling Lead and Zinc Concentrate

Lead concentrates are delivered to a distributor by a 4 inch Wilfley pump and are divided between three thickeners each 19 1/2 feet in diameter. The overflow from each thickener, which sometimes carries a thin film of froth, is delivered to a tank 14 feet in diameter and without peripheral overflow. A simple froth-retaining baffle catches any persistent froth and gives a clean overflow, which is pumped to the circuit water tanks.

Mine water has been found to leave a froth-cutting property that is utilized in the froth-breaking sprays.

Feed to the thickeners, allowing for spray dilution, ranges between 15 and 25 percent solids, the underflow being maintained at 70 percent solids.

The underflow of the thickeners discharges into two 14-foot-diameter by 7-foot high wooden-stave tanks provided with Goldfield-type agitators driven at 10 r.p.m. by 5-horsepower motors.

From these tanks concentrates are pumped at intervals to the smelter, each pumping consisting of 65 to 70 tons and requiring approximately 1 hour. A 3-inch Wilfley pump, belt-driven at 1,440 r.p.m. by a 30-horsepower motor, delivers the concentrates through a 3 inch pipe line to the smelter receiving tank. The pipe lines rises vertically to 10 feet above the pump and then falls uniformly to the discharge at a 3-1/2-percent grade. Its over-all length is 650 feet; a duplicate line is provided.

The power required by the pump falls slowly from full load (30 horsepower) at the beginning of pumping to approximately half load.

Further dewatering of the lead concentrates is accomplished in Dorrco filters installed at the smelter.

Thickening, Filtering and Drying Zinc Concentrates

Zinc concentrates are delivered by launder from the final cleaner cells to two 30-foot Dorr thickeners constructed of 10-foot wooden staves; type-R mechanism with overload alarm and relay is provided.

The peripheral launders originally provided on the thickeners have been removed owing to the difficulty experienced in breaking down the froth even with mine-water sprays. The thickeners are now operated in series, a simple froth baffle protecting a draw-off pipe on one side of the first thickener, the outflow from which, faintly turbid, is clarified by passing across the second thickener. Formerly, cloudy overflows were general, due to broken froth overflowing, but no difficulty is now experienced with solids in suspension.

Feed to the thickeners, including dilution by sprays, is between 20 and 30 percent solids, the underflows being maintained at 70 percent solids. The underflows are collected in a 14-foot-diameter by 7-foot wooden-stave tank similar to those used for the lead concentrates and pumped at intervals to the zinc-concentrate filter plant by a 4-inch Wilfley pump V-rope-driven at 1,470 r.p.m. by a 25-horsepower motor that is fully loaded during pumping.

The pipe line is 3 inches in diameter and 1,300 feet long, the receiving tank at the filter plant being 60 feet below the pump; the pipe line rises to 23 feet above the pump for the first 300 feet and then falls steadily to the outlet.

The zinc filter plant is provided with two storage tanks, each 12 feet in diameter by 12 feet in height and of wooden-stave construction (fig. 16). Agitator gear is similar to that in the lead-concentrate tanks.

These tanks feed by gravity to one 6-lead, 8 1/2-foot-diameter, American-type filter, a 3-inch Wilfley being provided to return overflow and drainage to the storage tanks. The filtering area is 560 square feet; the present output of 160 tons daily is handled with ease in two 8-hour shifts.

Roller discharges were tried on the filters but were abandoned in favor of curved-blade scrapers. These are now provided with replaceable molded-rubber edges that have reduced the wear on bags considerably.

Bags are made of 17-1/2-ounce cloth, their life averaging 80 operating days. They are generally patched several times before being discarded.

The filtrate is discharged into a separator, from which it is withdrawn by a centrifugal pump driven at 1,400 r.p.m. through V ropes by a 2-horsepower motor. The vacuum pump is connected to the separator via barometric leg and moisture trap. It is a 20- by 10-inch double-acting dry pump driven at 250 r.p.m. by a 40-horsepower motor requiring 30 to 35 horsepower and provides 22 to 24 inches of vacuum under normal operating conditions.

The filter cake, averaging 12 percent H_2O with present concentrates, discharges to a 30-inch belt conveyor that delivers either to the drier or to a belt system leading to a radial boom stacker-conveyor covering a stock pile.

The drier is a 7-foot-diameter by a 40-foot-long parallel-flow, direct-fired unit driven at 6.95 r.p.m. by a 50-horsepower motor through V ropes, speed reducer, and train of gears. It is inclined at 3/4-inch per foot from feed to discharge end. Six rows of baffles project inward from the shell and lift the concentrates into the stream of hot gases. Under normal running the drier requires 20 horsepower.

Coal is burned in a firebox of 36 square-feet area, draft being provided by a fan with a capacity of 13,000 cubic feet per minute at 4-inch water gage and 50° F. at 960 r.p.m. The fan is directly driven by a 20-horsepower motor and under normal conditions requires 12 horsepower handling 7,100 cubic feet per minute at 150° F., although these conditions vary as volume is varied to suit the feed rate.

Fuel required per ton of concentrates, inclusive of daily starting-up losses, ranges up to 30 pounds, according to the type of concentrate and season of the year. Moisture evaporated per pound of coal averages between 4 and 6 pounds.

The concentrates are discharged from the drier at 6-percent moisture and are delivered by 18-inch belt conveyor directly into railway cars.

Concentrates are reclaimed from the stock pile when required, either by scraper loaders or a steam shovel.

Dewatering and Disposal of Tailings

The tailings from the three rows of zinc-rougher cells flow by gravity through a launder and pipe line to a pair of 6-inch Wilfley pumps, each connected to a 12 inch wood-stave pipe line and belt driven by a 75-horsepower motor. One pump handles the flow and delivers it to an elevated launder system, where the tailings flow by gravity to two tailings dams, each formed by damming a conveniently situated disposal area. Only one dam is in regular use, the other being held in reserve and used for a storage basin for return water.

A launder is carried across the face of the dam, and tailings are distributed through holes in the launder bottom, the face being carried up to a 1 : 1 slope until the top of the dam approaches the level of the launder, which is then lifted 10 feet and set back 30 feet from the face of the dam and the cycle repeated. Each lift lasts approximately 12 months.

With sulfide ores, the dam wall is firm and has given no trouble with breakways, but the lighter carbonate ores gave trouble on occasions following heavy rain, water-logged areas sloughing away from the wall.

The water and slimes drain away from the face to form a clean pool; one dam is mainly used as a storage basin, the water being reclaimed for mill use by pontoon pumps, each having a capacity of 1,000 gallons per minute at 160-foot head. A 75-horsepower motor is connected to the pump by V ropes; the load is varied to suit the volume of water required at the mill.

Sampling

Mill heads are sampled at the overflow from the primary classifiers by automatic samplers, lead concentrates and final tailings also being sampled automatically. Zinc concentrates and intermediate products are hand-sampled at half-hourly intervals by a shift sampler.

A check determination of lead-concentrate assay and tonnage is made at the smelter, where accurate volume measurements and weight per unit-volume determinations enable the input tonnage to be calculated.

Mill pulp samples are filtered in pressure filters and delivered to the sample drying and bucking room.

Calculated mill output and smelter intake figures generally check closely.

Weightometers, on which mill tonnage is based, are calibrated once a week.

Accurate composite samples are assayed at the end of each month for a balance, which is reconciled with the smelter figures for the same period. A daily mill balance also is reported.

Conventional three-product formulas are used for computing the recoveries.

Wages

Wage rates are fixed according to industrial awards and are varied by the State Industrial Arbitration Court. Rates that prevailed in June 1938 for a working period of 88 hours per fortnight are as follows:

	Rate per 8-hour shift
Crushing-plant operator	$4.00
Crushing-plant helper	3.83
Ball-mill operator	4.67
Flotation operator	4.67
Flotation helper	3.83
Tailings-dam operator	3.83
Zinc filter-plant operator	4.07
Sampler	3.83
Reagent mixers	3.83
Greasers	3.83
Repairman-mechanic	4.80
Pipe fitter	4.13
Rigger and scraper driver	4.33
Millwright-carpenter	4.80
Shift electrician	5.20
Electrician	4.63
Repair helper	3.70
Laborers, not classified	3.70

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Mill Production - Operating Data and Costs

Table 2 gives essential yearly operating data and metallurgical results obtained for 1935, 1936, and 1937.

Milling costs per ton of ore milled for the production of lead concentrates are shown in table 3.

Zinc concentrate is considered as a byproduct, and only actual production cost is allocated. Table 4 gives the cost segregation for the production of zinc concentrate.

In the following tables, present milling results of a typical month's operation (May 1938) are given.

Table 5 shows the screen analyses of various mill products and assays of mill key products.

The assay screen analyses of mill products are given in table 6.

Miscellaneous metallurgical and operating data are shown in table 7.

Table 8 shows current lead-milling cost segregation.

Table 9 is a summary of consumption of flotation reagents and grinding balls.

Table 10 shows power distribution and consumption.

Table 11 contains analyses of mill waters.

Table 12 is a summary of costs in units of labor.

TABLE 1. - Recovery of silver and lead in lead concentrates and zinc in zinc concentrates

Period	Recovery, percent			Remarks
	Lead concentrate		Zinc concentrate	
	Silver, percent	Lead, percent	Zinc, percent	
March to June 1935.....	71	75.5	-	Symons short-head crusher operating May 1935. Primary Dorr F classifiers in operation July 1935.
July to December 1935.	68.7	72.3	-	Equipment for recovery of zinc concentrates in operation October 1935.
January to June 1936.....	68.8	72.7	34.7	-
July to December 1936.	73.2	77.2	49.1	-
January to June 1937.....	74.6	78.1	46.1	Crushing-plant flow sheet revised January 1937. Dorr model D bowl classifiers replaced by Dorr QSFXB bowl classifiers March 1937.
uly to December 1937.	77.4	81.3	48.1	Tertiary grinding unit in operation December 1937.
anuary to June 1938.....	82.1	85.5	51.8	-

TABLE 2. - Mill production - operating data

			July 1935 June 1936	July 1936 June 1937	July 1937 June 1938
Heads	Tons milled		692,596	647,381	664,093
	Silver, oz.		5.5	6.1	6.5
	Lead, percent		7.8	8.3	8.9
	Zinc, percent		11.1	9.9	9.8
Lead concentrates	Tons produced		95,402	106,053	126,726
	Silver, oz.		27.8	27.3	27.2
	Lead, percent		41.2	39.4	39.1
	Zinc, percent		11.4	10.8	10.4
	Silver, oz.		2,657,229	2,899,202	3,455,368
	Lead, tons		39,296	41,773	49,562
	Zinc, tons		10,876	11,436	13,172
Zinc concentrates	Tons produced		52,459	58,468	61,807
	Silver, oz.		3.6	2.8	2.4
	Lead, percent		3.3	2.3	1.7
	Zinc, percent		50.8	52.1	52.7
	Silver, oz.		189,354	167,644	149,038
	Lead, tons		1,728	1,339	1,049
	Zinc, tons		26,673	30,449	32,568
Final tails	Silver, oz.		1.8	1.7	1.5
	Lead, percent		2.4	2.2	1.8
	Zinc, percent		7.2	4.6	4.1
Recoveries, percent	In lead concentrates	Ag	68.8	73.9	79.8
		Pb	72.4	77.6	83.5
	In zinc concentrates	Zn	34.7	47.6	49.9
Reagents, pounds per ton	Aerofloat 25		0.010	0.015	0.096
	Aerofloat 31		.175	.178	.029
	Cresylic acid		.004	.085	.117
	Xanthate		.287	.314	.336
	Soda ash		1.743	1.356	1.222
	Sodium sulfide		.303	-	-
	Zinc sulfate		.533	.576	.526
	Cyanide		1.051	.575	.526
	Copper sulfate		1.419	.923	.933
	Sodium aerofloat		.174	.056	.074
Grinding balls, pounds per ton			1.38	1.40	1.67
Total kw.-hr. per ton			28.847	32.747	34.056
Percent total running time			96.6	95.3	98.6

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TABLE 3. - Summary of costs, year ended June 30, 1938

	Tons
Ore treated	664,093
Lead concentrates produced	126,726
Zinc concentrates produced	61,807

Milling cost, in dollars per ton ore treated

	Labor	Supplies	All other direct milling costs	Total milling costs
Salaries and wages...	0.068	-	-	0.068
Crushing............	.042	0.063	-	.105
Grinding............	.019	.040	-	.059
Classifying.........	.003	.012	-	.015
Flotation...........	.024	.076	-	.100
Concentrate handling.	.008	.002	-	.010
Tailings disposal....	.013	.020	-	.033
Pumps...............	.002	.014	-	.016
Research............	.004	.001	-	.005
Power...............	-	-	0.188	.188
Assaying............	-	-	.026	.026
Sampling............	-	-	.013	.013
Water...............	-	-	.002	.002
Reagents............	-	-	.170	.170
Grinding balls.......	-	-	.063	.063
Miscellaneous........	-	-	.039	.039
	.183	.228	.501	.912

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TABLE 4. - Summary of costs, year ended June 30, 1938

	Tons
Ore treated	664,093
Lead concentrates produced	126,726
Zinc concentrates produced	61,807

Cost of production of Zn concentrates, in dollars

	Labor		Supplies		Other Costs		Total	
	Per ton ore treated	Per ton concentrate produced	Per ton ore treated	Per ton concentrate produced	Per ton ore treated	Per ton concentrate produced	Per ton ore treated	Per ton concentrate produced
Flotation thickening......	0.013	0.139	0.003	0.032	0.006	0.064	0.022	0.235
Filtering, drying stacking, loading	.020	.214	.005	.054	.011	.118	.036	.386
Power............	-	-	-	-	.040	.429	.040	.429
Reagents.........	-	-	-	-	.092	.986	.092	.986
Coal.............	-	-	-	-	.009	.096	.009	.096
Total............	-	-	-	-	-	-	.199	2.132

TABLE 5. - Screen analyses, May 1938

			Classifier overflow							
	Mill feed		Primary		Secondary		Tertiary		Final tails	
Screen mesh, inches	Percent weight	Percent cumulative	Percent weight	Percent cumulative	Per cent weight	Percent cumulative	Percent weight	Percent cumulative	Percent weight	Percent cumulative
+0.371	0.6	0.6	-	-	-	-	-	-	-	-
+3	15.3	15.9	-	-	-	-	-	-	-	-
+4	24.0	39.9	-	-	-	-	-	-	-	-
+6	12.0	51.9	-	-	-	-	-	-	-	-
+8	12.0	63.9	-	-	-	-	-	-	-	-
+10	6.3	70.2	2.5	2.5	-	-	-	-	-	-
+14	5.7	75.9	4.5	7.0	-	-	-	-	-	-
+20	3.2	79.1	5.0	12.0	-	-	-	-	-	-
+28	3.2	82.3	7.5	19.5	-	-	-	-	-	-
+35	2.9	85.2	9.5	29.0	-	-	-	-	-	-
+48	2.2	87.4	9.0	38.0	-	-	-	-	-	-
+65	1.5	88.9	8.0	46.0	2.5	2.5	-	-	-	-
+100	1.3	90.2	6.5	52.5	5.0	7.5	-	-	3.0	3.0
+150	1.0	91.2	5.5	58.0	8.5	16.0	6.0	6.0	8.5	11.5
+200	.8	92.0	4.0	62.0	9.0	25.0	8.5	14.5	11.5	23.0
-200	8.0	-	38.0	-	-	-	-	-	-	-
+325	-	-	-	-	10.0	35.0	9.5	24.0	13.0	36.0
-325	-	-	-	-	65.0	-	76.0	-	64.0	-

Assays of mill products for May 1938

	Ag, oz.	Pb, percent	Zn, percent	Fe, percent	S percent	Insoluble, percent	Cu, percent
Classifier overflow	7.4	9.2	9.3	11.0	13.8	35.8	0.1
Lead concentrates..	30.6	39.5	10.0	14.3	26.0	6.0	.35
Lead tails.........	1.7	1.7	9.1	9.7	11.0	43.2	.06
Zinc concentrates..	2.7	1.7	52.8	7.5	31.7	3.4	.1
Final tails........	1.6	1.7	3.5	9.8	8.4	48.6	.05

TABLE 6. - Screen analyses and assays of mill products, May 1938

Flotation feed (tertiary classifier overflow)

Screen mesh	Percent weight	Percent cumulative	Assays, percent Pb	Zn	Fe	Distribution, percent Pb	Zn	Fe
+ 150	6.0	6.0	1.8	1.5	4.4	1.2	1.0	2.4
+ 200	8.5	14.5	2.8	2.7	3.8	2.6	2.5	2.9
+ 325	9.5	24.0	4.7	10.1	11.6	4.9	10.3	10.0
- 325	76.0	-	11.0	10.5	12.2	91.3	86.2	84.7
	100.0		9.2	9.3	11.0	100.0	100.0	100.0

Lead final concentrate

Screen mesh	Percent weight	Percent cumulative	Assays, percent Pb	Zn	Fe	Distribution, percent Pb	Zn	Fe
+ 200	-	-	-	-	-	-	-	-
+325	5.5	5.5	29.8	18.4	10.2	4.2	10.1	3.9
-325	94.5	-	40.0	9.5	14.6	95.8	89.9	96.1
	100.0		39.5	10.0	14.3	100.0	100.0	100.0

Lead cleaner tail

Screen mesh	Percent weight	Percent cumulative	Assays, percent Pb	Zn	Fe	Distribution, percent Pb	Zn	Fe
+ 200	6.0	6.0	9.0	8.6	7.8	6.3	3.5	2.7
+ 325	10.0	16.0	12.5	16.3	12.2	14.7	10.9	7.0
- 325	84.0	-	8.0	15.2	18.7	79.0	85.6	90.3
	100.0		8.5	14.9	17.4	100.0	100.0	100.0

Lead middling

Screen mesh	Percent weight	Percent cumulative	Assays, percent Pb	Zn	Fe	Distribution, percent Pb	Zn	Fe
+ 150	-	-	-	-	-	-	-	-
+ 200	4.0	4.0	12.4	12.5	9.4	8.0	3.0	2.2
+ 325	10.0	14.0	11.3	17.7	14.5	18.3	10.7	8.5
- 325	86.0	-	5.3	16.6	17.8	73.7	86.3	89.3
	100.0		6.2	16.5	17.1	100.0	100.0	100.0

Lead tail

Screen mesh	Percent weight	Percent cumulative	Assays, percent Pb	Zn	Fe	Distribution, percent Pb	Zn	Fe
+ 100	-	-	-	-	-	-	-	-
+ 150	11.5	11.5	1.7	2.5	5.0	11.5	3.2	5.9
+ 200	12.0	23.5	2.65	5.5	8.0	18.8	7.2	9.8
+ 325	14.0	37.5	2.6	8.4	11.9	21.5	12.9	17.2
- 325	62.5	-	1.3	11.2	10.5	48.2	76.7	67.0
	100.0		1.7	9.1	9.7	100.0	100.0	100.0

TABLE 6. - Screen analyses and assays of mill products, May 1938 - (Continued)

Zinc concentrate

creen esh	Percent weight	Percent cumulative	Assays, percent			Distribution, percent		
			Pb	Zn	Fe	Pb	Zn	Fe
150	-	-	-	-	-	-	-	-
200	3.0	3.0	3.65	44.6	8.5	6.4	2.5	3.4
325	11.5	14.5	3.15	46.5	9.1	21.3	10.1	14.0
325	85.5	-	1.44	53.9	7.2	72.3	87.4	82.6
	100.0		1.7	52.8	7.5	100.0	100.0	100.0

Zinc cleaner tail

creen esh	Percent weight	Percent cumulative	Assays, percent			Distribution, percent		
			Pb	Zn	Fe	Pb	Zn	Fe
150	2.5	2.5	3.6	31.9	6.9	2.7	2.1	1.7
200	12.5	15.0	4.2	40.1	8.4	15.7	13.0	10.6
325	24.0	39.0	4.2	39.4	9.6	30.3	24.4	23.2
325	61.0	-	2.8	38.4	10.5	51.3	60.5	64.5
	100.0		3.3	38.7	9.9	100.0	100.0	100.0

Final tail

creen esh	Percent weight	Percent cumulative	Assays, percent			Distribution, percent		
			Pb	Zn	Fe	Pb	Zn	Fe
100	3.0	3.0	1.2	1.9	3.9	2.1	1.6	1.2
150	8.5	11.5	1.7	2.8	5.1	8.4	6.8	4.4
200	11.5	23.0	2.35	4.4	7.8	15.6	14.5	9.1
325	13.0	36.0	2.45	4.85	12.1	18.4	18.0	16.1
325	64.0	-	1.5	3.22	10.6	55.5	59.1	69.2
	100.0		1.7	3.5	9.8	100.0	100.0	100.0

TABLE 7. - Metallurgical Data for May 1938

Total tons ore treated	55,216
Days operated	30
Hours operated per day	24
Average tonnage per 24 hours	1,840
Average sections running	3
Average tonnage per 24 hours per section	613
Total tons concentrate (lead plus zinc)	15,960
Total average tons concentrates per 24 hours	532
Total tons lead concentrates	10,923
Average tons lead concentrates per 24 hours	364
Total tons zinc concentrates	5,037
Average tons zinc concentrate per 24 hours	168
Recovery of silver in lead concentrates, percent	81.5
Recovery of lead in lead concentrates, percent	85.0
Recovery of zinc in zinc concentrates, percent	51.9
Ratio of concentration lead section, tons of original ore into 1	5.06
Ratio of concentration zinc section, tons of original ore into 1	10.96
Ball consumption, pounds per ton ore	1.35

TABLE 8. - Summary of Costs, May 1938

	Tons
Ore treated	55,216
Lead concentrates produced	10,923
Zinc concentrates produced	5,037

Milling costs, in dollars per ton ore treated[1]

	Labor	Supplies	All other direct milling costs	Total milling cost
Salaries and wages...	0.065	-	-	0.065
Crushing.............	.036	.057	-	.093
Grinding.............	.020	.043	-	.063
Classifying..........	.002	.006	-	.008
Flotation............	.023	.078	-	.101
Concentrate handling.	.008	.001	-	.009
Tailings disposal....	.014	.017	-	.031
Pumps................	.001	.009	-	.010
Research.............	.004	.003	-	.007
Power................	-	-	0.169	.169
Assaying.............	-	-	.026	.026
Sampling.............	-	-	.012	.012
Water................	-	-	.002	.002
Grinding balls.......	-	-	.069	.069
Reagents.............	-	-	.161	.161
Miscellaneous........	-	-	.029	.029
	.173	.214	.468	.855

[1] These costs are for lead flotation only. Zinc concentrates are treated as a byproduct and bear only production cost from tails of lead flotation circuit.

TABLE 9. - Summary of flotation reagents and grinding balls, May 1938

Flotation reagents

	Reagent	Pounds per ton of ore treated
Lead flotation	Soda ash	1.14
	Cyanide	.53
	Zinc sulfate	.53
	Xanthate	.25
	Aerofloat 31	.005
	Aerofloat 25	.08
	Cresylic acid	.07
Zinc flotation	Copper sulfate	.90
	Sodium aerofloat	.07
	Xanthate	.07
	Aerofloat 31	.005
	Cresylic acid	.01

Grinding balls

Description	Pounds per ton of ore treated
4-inch forged chrome-steel balls..................	0.68
3-inch forged chrome-steel balls.................	.04
2-inch cast white alloy iron balls...............	.84
1-1/2-inch white alloy cast-iron balls...........	.29
	1.85

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TABLE 10. - Power distribution, November 1938

	Kw.-hr. per ton of ore treated	Distribution, percent
Crushing plant:		
Crushers..........................	1.31	4.09
Screens, conveyors, etc.	.78	2.44
	2.09	6.53
Concentrator:		
Primary grinding and classification............................	7.35	22.96
Secondary grinding and classification..........................	7.74	24.18
Tertiary grinding and classification...........................	2.70	8.43
	17.79	55.57
Lead flotation..................	5.72	17.87
Lead-concentrate handling.......	.25	.78
	5.97	18.65
Zinc flotation..................	3.46	10.81
Zinc concentrate handling.......	.16	.50
Zinc concentrate filtering and drying..........................	.34	1.06
	3.96	12.37
Tailings disposal and return water circuit..................	1.75	5.47
Lighting........................	.45	1.41
Total kw.-hrs. per ton ore treated...	32.01	100.00

TABLE 11. - Analysis of concentrator water

	Parts per million		
	Mill-circuit water	Mine water	Rifle Creek Reservoir
Total solids.....	3,145	3,244	76.0
Volatile and organic matter..	83	254	-
Fixed solids.....	3,062	-	-
CaO..............	222	410	13.0
MgO..............	250	304	6.0
Na_2O.............	890	582	9.7
Fe...............	4	2.8	.87
Mn...............	-	-	.21
SiO_2.............	18	24	8.0
Cl...............	545	490	2.6
SO_3..............	1,142	1,231	Nil

TABLE 12. - Summary of costs in units of labor, May 1938

	Tons
Ore treated	55,216
Lead concentrates produced	10,923
Zinc concentrates produced	5,037

	Number men employed per day	Man-hours per ton ore treated
Foreman	3	0.013
Crushing plant	1/17	.061
Grinding:		
Primary and secondary	6	.026
Tertiary	3	.013
Flotation:		
Lead	3	.013
Zinc	3	.013
Handling:		
Lead concentrate	3	.013
Zinc concentrate	3	.013
Zinc concentrate filtering	5	.021
Tailings disposal	7	.030
Greasing	4	.017
Sampling	5	.021
Reagent mixing	2/2	.008
Maintenance:		
Mechanical	2/15	-
Electrical	2/ 8	-
Total	87	
Total operating man hours/ton ore treated		.262

1/ Operating 6 days per week.
2/ Operating 5-1/2 days per week.